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# PROGRESS REPORT - BACTERIALLY ASSISTED LEACHING OF FALCONBRIDGE COPPER ORE

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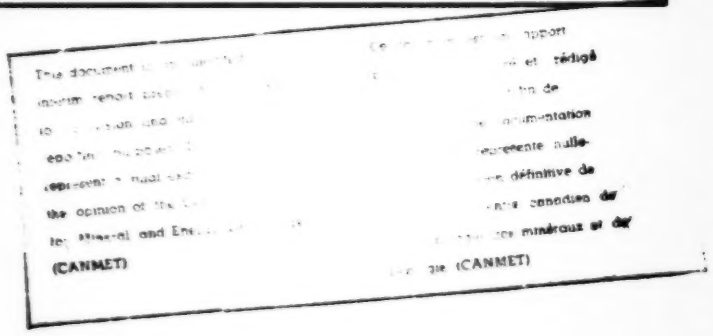
August 1984



Project MRP-3.4.0.2.12
Microbiological Leaching

35 pp

MINERALS RESEARCH PROGRAM
MINERAL SCIENCES LABORATORY
DIVISION REPORT MRP/MSL 84-92 (IR)

NOV 22 1984
MICROMEDIA

# PROGRESS REPORT - BACTERIALLY ASSISTED LEACHING OF FALCONBRIDGE COPPER ORE

by

V. Sanmugasunderam and S. Rajan

## SUMMARY

Low grade copper ore from the Lake Opemiska mine of Falconbridge Mines Limited was leached by ferric sulphate at a pH between 1.5 and 2.0. The leaching rate, which was independent of the ferric sulphate concentration, increased with time and accelerated to more than five times the initial rate after 140 days. Consumption of acid during leaching was 0.18 mg per kg of ore per day. The leaching rate after a period of 140 days is 1.1 per cent per month. When compared to those leaching rates found in the literature for which economic evaluations have been done, the rates obtained with the Opemiska ore appear to be promising.

# RAPPORT SUR L'ACTIVITÉ EN COURS-LIXIVIATION BACTÉRIENNE DU MINERAI DE CUIVRE DE LA FALCONBRIDGE

par

V. Sanmugasunderam et S. Rajan

## RÉSUMÉ

On a lixivié du minerai à faible teneur en cuivre de la mine du lac Opemiska de la Falconbridge Limited à l'aide de sulfate ferrique à un pH de 1,5 à 2,0. La vitesse de lixiviation qui était indépendante de la concentration de sulfate ferrique s'est accrue avec le temps et s'est accélérée à plus de cinq fois la vitesse initiale après 140 jours. La consommation d'acide durant la lixiviation a été de 0,18 mg par kg de minerai par jour. La vitesse de lixiviation après une période de 140 jours était de 1,1 p. 100 par mois. Si on compare ces résultats aux résultats mentionnés dans les documents scientifiques et pour lesquels des appréciations économiques ont été faites, les résultats de lixiviation obtenus avec le minerai de la mine du lac Opemiska sont prometteurs.

# INTRODUCTION

One project in CANMET's (Canada Centre for Mineral and Energy Technology) Mineral Processing Technology program has the objective "to develop biometallurgical processes to extract and recover residual metal values from sulphide ores, specifically those portions of ore deposits that are usually left underground". The utilization of this ore, usually left as pillars or below cut-off grade, could extend the productive life of mines nearing the exhaustion of their ore reserves. The biometallurgical process could also be used as an economically attractive adjunct to conventional mining operations.

Extraction of metals by the use of microorganisms is no longer considered to be a novel process. In natural weathering, microorganisms play an important role in decomposing minerals; man has simply utilized this microbial action to assist in leaching metals from ores. There are numerous examples of bacterially-assisted processes - in copper heap and dump leaching, and in uranium recovery procedures - being used in commercial operations.

The objectives of CANMET's project is to expand the use of underground in-place leaching in the base metals mines in Canada. The process, shown in the conceptual flowsheet in Fig. 1, consists of blasting the ore underground into approximately minus 15 cm (minus 6 in.) size, pumping leach solution into bulkheaded stopes, retaining the liquor for the necessary length of time, and then withdrawing the pregnant metal-containing solution. The pregnant liquor is then purified and concentrated by means of ion-exchange or solvent extraction processing, and the metal then removed from solution by suitable techniques such as precipitation, cementation or electrowinning.

The evolution of a biometallurgical process consists of a succession of stages; the investigation may be terminated at any stage should the data indicate the impracticability of the process, or the unsuitability of the ore. These stages are:

## Stage I Amenability Testing

Representative samples of the ore will be chemically and mineralogically analyzed, and subjected to amenability testing which includes testing for porosity and bench scale bacterial leaching tests to

determine the per cent extraction of the metals. These bench-scale tests will be done on finely ground ore in "shake-flask" tests.

The results of these amenability tests will show whether or not the ore is suitable for bacterial leaching.

Stage II Mini-Plant Study

A. Mini-plant column leaching

Half tonne of minus 15 cm (minus 6 in.) ore will be flood (or percolation) leached under varying operating conditions, for the purpose of developing a mathematical model which could be used to predict the leaching characteristics on the large scale.

B. Feasibility study - underground operations

(1) Mine-site Selection

The suitability of the mine, for underground leaching, will be evaluated using a logical sequence of selection criteria including geology, accessibility, mine configuration, ore grade, tonnage and available facilities.

(2) Pre-feasibility of Underground Operations

A blasting design for both trickle and flood leaching will be developed and costed. Appropriate systems for acid storage, bulk-heading solution distribution and collection, ventilation, etc., will be planned and costed.

(3) Pre-feasibility of Metallurgical Circuit

Preliminary flow sheet, to produce a marketable end product, including material and heat balances will be drawn up. Typical equipment needed will be sized and costed.

(4) Overall Assessment and Sensitivity Analysis

A complete schema with capital and operating costs will be detailed. Cash flow analysis and sensitivity analyses on major cost factors will be determined and other economic criteria identified.

These pre-feasibility studies will enable the economic extraction rate to be stipulated.

C. Feasibility study - solution treatment and metal recovery

A study of alternative methods of treatment of the pregnant solution with respect to purification, concentration and recovery of the metals will be conducted to formulate an estimate of the cost of this operation and thus ascertain its feasibility.

Stage III On-Site Pilot Study

An on-site pilot study, on a scale determined by the subject mining company, will be conducted to generate the data necessary for the scale-up to industrial scale operations.

Stage IV On-Site Demonstration

An on-site demonstration of the complete process will be conducted to demonstrate the economic viability of the biometallurgical extraction process.

## DETAILED DESCRIPTION OF THE IN-PLACE LEACHING PROCESS

The underground in-place bacterial leaching process for sulphide ores, proposed by CANMET, Energy, Mines and Resources Canada, consists of a series of steps involving leaching, metal solution purification, and recovery of the metals from solution as shown in the flowsheets in Fig. 1 and 2. Intended primarily for the treatment of below cut-off grade ore (normally left underground), the operating costs must be kept to a minimum to enable the process to be economically viable.

Comminution costs will be kept to a minimum by blasting the ore in-place to minus 15 cm (minus 6 in.), with no further crushing or grinding. The stope, prior to blasting will have been prepared so that in the ensuing leaching step the solution will be retained in a collection pit (sealed to prevent run-off loss of the solution).

Spraying of the blasted ore will be done by means of simple lawn-sprinkler type of apparatus; the leaching solution will trickle through the rubbled heap and collect in the receiving pit. The leaching solution, essentially ferric sulphate acidified with sulphuric acid to pH 1.5 and containing Thiobacillus ferrooxidans bacteria, will be added at a rate (determined in previous mini-plant column leaching tests) such that

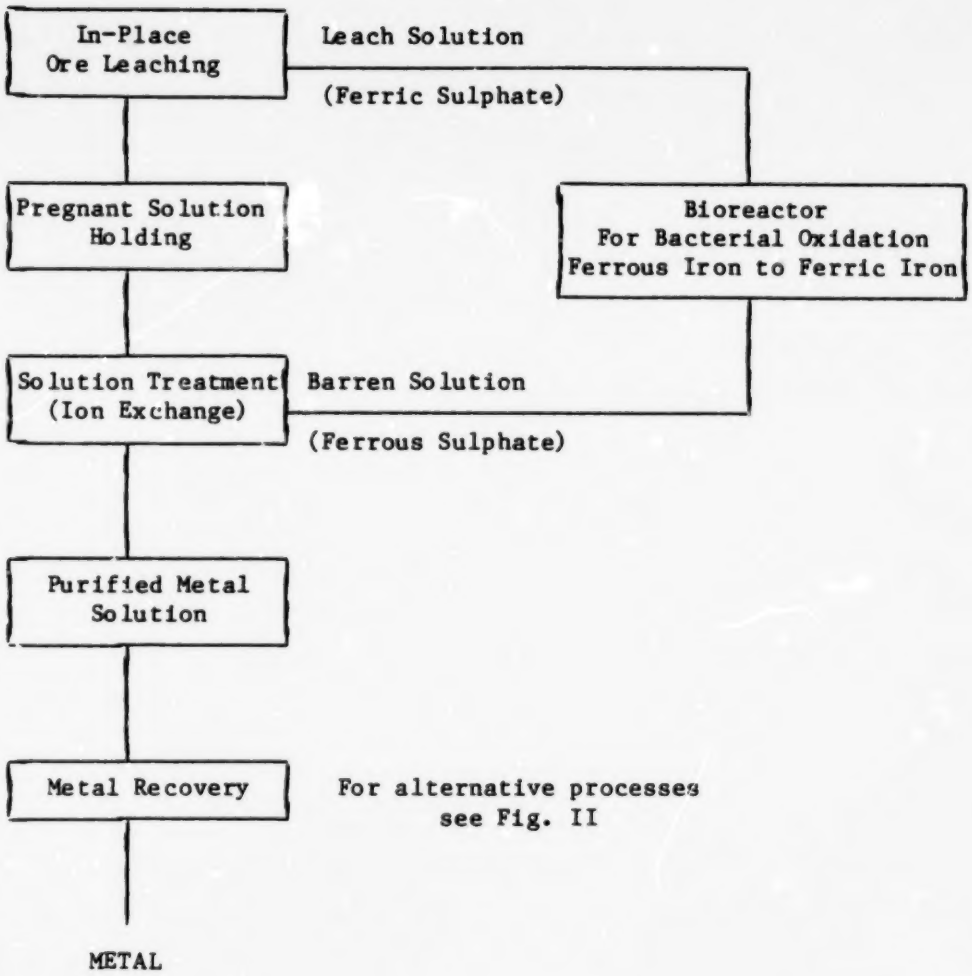
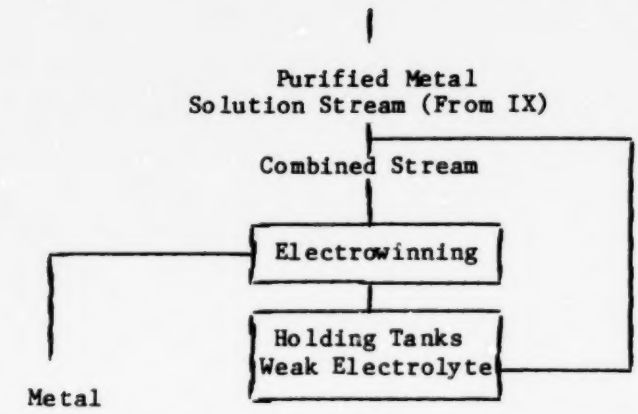
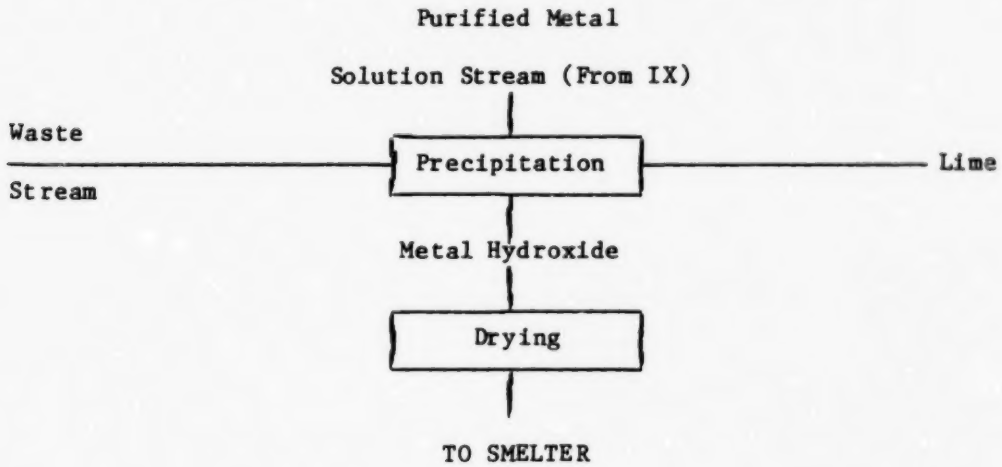


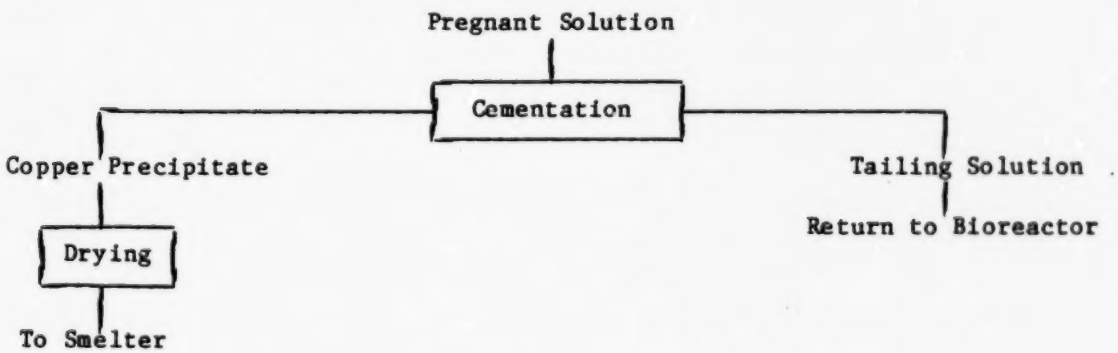

Fig. 1 - Block Diagram for Bacterial Leaching




Alternative (a):- Metal Recovery by Electrowinning




Alternative (b):- Metal Recovery by Precipitation




Alternative (c):- Metal Recovery by Cementation

Fig. 2. Block Diagram for Metal Recovery

the ferric iron content of the solution is reduced to ferrous during the leaching action, according to the reaction

$$Fe_2(SO_4)_3 + MS \longrightarrow 2FeSO_4 + MSO_4 + S^0$$

(Mineral Sulphide)

The optimum concentration of the ferric iron will be determined based on the results of the mini-plant experiment.

This pregnant solution will pass to a holding tank and then be pumped to the ion exchange treatment plant. By passage through the appropriate ion-exchange resin, the metal values will be absorbed onto the resin (with the ferric iron) and the barren solution will pass to a holding tank.

Lime slurry will be added to this barren solution (containing ferrous ion) until a value of pH 2.5 is reached, and the adjusted solution will be pumped to a bioreactor where the ferrous iron will be oxidized to ferric iron by the action of T. ferrooxidans bacteria. This adjustment to pH 2.5 is necessary to provide the optimum range (pH 2.0-3.0) for the growth of the bacteria.

Upon completion of the oxidation of ferrous iron to the ferric state, the solution is acidified to pH 1.5 by the addition of sulphuric acid. By lowering the pH value, the precipitation of jarosite and its deposition onto the surface of the ore is avoided. (Such deposition would clog the pores of the ore and thus hinder the access of the solution to the metals).

In this recycling loop of the leach solution, small amounts of nutrients (phosphate, ammonium sulphate, and trace quantities of calcium nitrate and magnesium sulphate) necessary for growth of the bacteria will be added to the bioreactor. A bleed of the barren solution will occur prior to pH adjustment to accommodate the solution balance (water being added during the pH adjustment and nutrient addition) and also will reduce the build-up of impurities such as sulphate, etc.

To recover the metal values which have been leached from the ore, the ion-exchange resin is eluted with sodium chloride brine. The metals, which have varying desorption properties, elute according to the amount of brine added. Zinc and copper (two of the major metals of interest) are stripped simultaneously. The eluant solution contains other metal impurities, and further processing may be needed according to the ultimate recovery

procedure. For example, purification of the eluate by solvent extraction is necessary prior to electrowinning copper.

The detailed treatment of the eluant solution emerging from the ion exchange stage will be governed by the metals which are to be recovered and by the impurity content.

In the event of ion-exchange process being unsuitable, solvent extraction or cementation could be used for the recovery of the metal values.

## MATERIALS AND METHODS

The tests for determining the amenability of an ore to bacterial leaching include mineralogical and chemical analyses, immersion porosity tests and shake flask bacterial leaching tests.

To ensure that the ore was well mixed it was poured from the drums to a pile shaped in the form of a cone. The cone was flattened and divided into four parts. Each part was piled into a cone shape. Shovelfuls of ore from each of the four cones were taken sequentially and piled onto a single cone. This process of coning and quartering was carried out three times. The blended ore was screened into different size fractions and each fraction was weighed. Particles smaller than 0.4 cm ($\frac{1}{4}$ in.) were not used for mini-plant leaching.

Selected rocks were sent for mineralogical analysis and determination of porosity. Representative samples of the rock, fines and composite were ground to minus 106 µm and sent for chemical analysis.

The bacteria (iron-oxidizing bacteria) used for bacterial leaching were isolated from the mine waters of Elliot Lake area and were capable of metabolic activity at pH 1.0, although at a much lower rate than at the optimum pH.

### A. CHEMICAL ANALYSIS

Head samples of the ore were analysed using the wet and fire assay technique to determine the chemical composition of selected elements and radicals.

B. POROSITY

The porosity of a rock is usually determined by evacuating moisture and gases by vacuum and then forcing mercury into it under pressure. The pores intruded by this method are connecting pores and storage pores. Storage pores are sub-divided into micro, intermediate, nano and subnano pores.

Immersion porosity ($\Phi_I$) is defined by

$$\Phi_I = V_p / V_R \qquad \text{(Eq.1)}$$

where,

$V_p$ = volume of pore space

and $V_R$ = bulk volume of rock specimen containing mineral grains and pores.

The pore volume, $V_p$, is derived from taking the difference between the wet and dry weight of the specimen.

$$V_p = (W_w - W_d) / \delta_w \qquad \text{(Eq. 2)}$$

where,

$W_w$ = weight of specimen when saturated with water

$W_d$ = weight of specimen after oven drying

and $\delta_w$ = bulk density of water.

$$V_R = W_r / \delta \qquad \text{(Eq. 3)}$$

where,

$W_r$ = weight of the specimen at room dry condition

and $\delta$ = bulk density of rock in g/mL.

The immersion porosity measurements were carried out at the Electrical Rock Property Laboratory, Geological Survey of Canada.

Method of Investigation

Half disc specimens of 4.5 cm in diameter and 1.0 cm in thickness were used. First, the rock specimens were vacuum saturated with distilled water. Then the surfaces were dried with Kleenex and were weighed. The specimens were then placed in an oven at 110°C for 4 hours and weighed. The laboratory humidity was maintained at 40%.

Immersion porosity measured the pore volumes of interconnected pores and storage pores connected to interconnected pores. The pore volume as determined by immersion porosity contributes to ion-transport in the rock mass (1). Hence it is considered that immersion porosity is a better measure of the accessibility of metal values in the rocks than porosity measurements using the porosimeter.

Sedimentary rocks have a high immersion porosity, ranging from 5 - 15%. Tighter sedimentary rocks have an immersion porosity around 2%. The average value for immersion porosity, based on 200 determinations made on different crystalline rocks from Ontario, is 0.3 - 0.4% (2).

C. MINERALOGICAL ANALYSIS

Mineralogical investigation was conducted to obtain background information.

Method of Investigation

Polished sections were prepared and examined under the microscope to identify metallic minerals. The minerals present were also identified using the X-ray diffractometer. Ratios and particle size distribution of the metallic minerals were determined using the 'Quantimet 720 Image Analyser'.

D. SHAKE FLASK BACTERIAL LEACHING TEST

Samples of the ore crushed and pulverized to minus 75 μm were used for bacterial leaching. Five per cent (w/v) slurry of the ore made up with modified 9 K medium (3) was equilibriated to a pH of 2.5 and inoculated with T. ferrooxidans. The composition of modified 9 K medium is given in Table 1. The maximum amount of metal values solubilized is determined using atomic absorbtion spectrophotometer and the maximum possible biological extraction of the metal values calculated. This calculated value will give an indication of what percentage of the metal value in the ore could be expected to be leached out in practice.

Silverman and Lundgren (3) found that T. ferrooxidans was active in the pH range of 1.5 to 5.0. Optimum pH for oxidation of chalcopyrite (4), zinc sulphide (5), ferrous iron (6) cobalt sulphide (7), chalcocite and covellite (8), galena (9) and pyrite (10) was observed to be between 1.0 and 2.5. Although extraction rate varies with pH the end products are the same and hence, based on thermodynamics, it is postulated that the maximum amount that could be leached does not vary with pH.

E. MINI-PLANT COLUMN LEACHING

Ore fractions greater than 0.4 cm (¼ in.) were loaded into two polyvinyl chloride (PVC) columns 33 cm (13 in.) in diameter and 5.4 M (18 ft) tall. The fines which were only 3.75% of the total weight of ore received were left out to prevent plugging of the screen at the bottom of the columns. Each column was loaded with 417 kg (917.5 lb) of ore. The size distributions of the ore in the columns are in Table 2. The columns were filled with 160 L of pH 1.5 water. The water was recirculated and the pH values monitored using a pH stat until the pH remained constant for 2 days. When the pH stabilized, the water was drained out and a sample sent for analysis. The amount of copper dissolved by pH 1.5 water was only $3 \times 10^{-4}$ per cent and the concentration of Cu in the water was only 8 mg/L. After draining the water, each column was filled with 170 L of lixivant from the bioreactor. The principal constituent of the lixiviant was ferric sulphate (containing 5 g/L of Fe). Each column was loaded with 416 kg (917.5 lb) of ore. The lixiviant in one column (column 3) was recirculated and the pH values monitored using a pH stat, while the other column remained flooded. The flooded column was emptied every week for sampling purposes. An estimated quantity of acid required to maintain the pH below

1.8 was added to the flooded column daily and the pH adjusted to 1.5 weekly, when all the lixiviant was drained out.

Samples were withdrawn from both columns weekly and analyzed for copper, ferrous iron and total iron. The concentration of copper was determined by atomic absorption spectroscopy. Ferrous and total iron concentrations were determined by volumetric methods.

Since the difference in the leaching rate between the circulating column and the stationary column was very small, the circulation of the lixiviant was discontinued after day 134. Column 3, which was stationary, was drained on day 138 and a second experiment with fresh lixiviant was started in this column.

## LEACHING KINETICS

A kinetic model that would describe and predict the noncatalytic reaction of solid particles with surrounding fluid, as in the case of leaching ore particles by ferric sulphate lixiviant, is the shrinking core model (11). The shrinking core model has been applied to ore leaching by Madsen et al. (12), Braun et al. (13) and Meixner (14) with satisfactory results. The shrinking core model is based on the assumption that the reaction occurs first at the outer skin of the particle and then progresses into the solid leaving behind completely reacted material and inert solid leaching residue. Figure 3 represents an idealized ore particle showing the reaction zone of thickness (8). The original average radius is $r_{io}$ and the radius at time t is $r_i$.

As the zone moves to smaller values of $r_i$, unreacted mineral particles are included in the zone. At the trailing edge of the zone, the last vestige of unreacted mineral disappears. The mathematical analysis of the model follows the development presented by the workers cited earlier (12,13,14).

The rate of reaction within the reaction zone is given by the equation:

$$\left[\frac{dn}{dt}\right]_i = -\left[\frac{4\pi r_{io}^2}{\phi_{io}}\right] .\delta.n_p.A_p.C_i.k \qquad \text{(Eq. 4)}$$

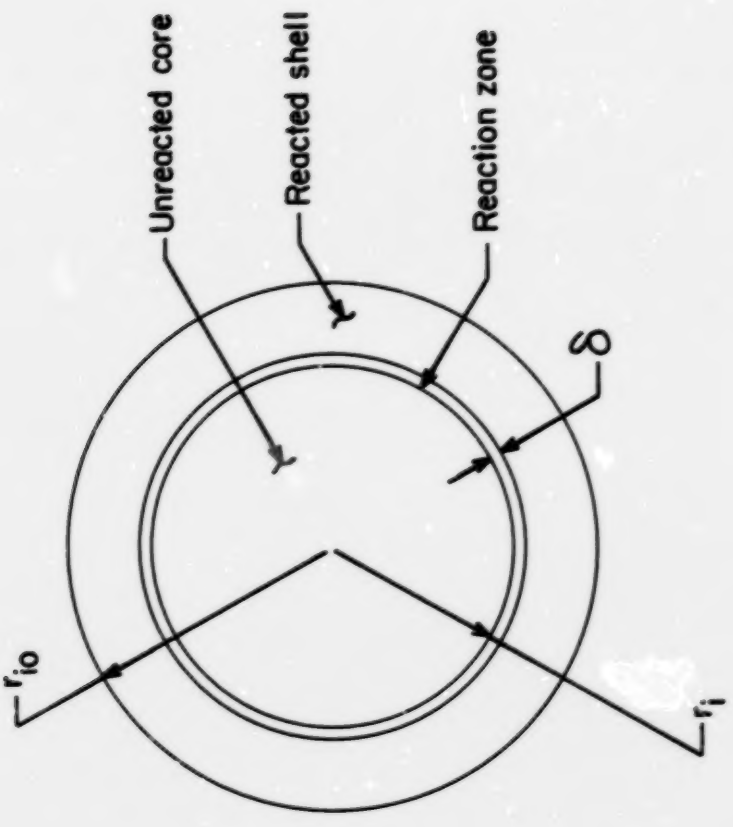



FIGURE 3 - Ore Particle of Radius $r_{io}$ Showing Reaction Zone.

where,

$k$ is the reaction rate constant

$C_i$ is the reactant concentration in the reaction zone

$A_p$ is the average area per sulphide particle

$n_p$ is the number of sulphide particles per unit volume

$\phi_{io}$ is a geometric factor, which corrects for boundary roughness and non-sphericity

and $n$ is the number of moles of unreacted copper minerals in the ore particle.

The diffusion through pores to the reaction zone, at steady-state is expressed by the equation:

$$\left[\frac{dn}{dt}\right]_i = -\left[\frac{4\pi r_i^2}{\phi_{io}}\right]\left[\frac{Df}{\sigma}\right]\left[\frac{dC}{dr}\right]_i = J_i \qquad \text{(Eq. 5)}$$

where,

D.f is the effective diffusivity of the reagent

$\sigma$ is the stoichiometry number - the number of moles of oxidant

and $f$ accounts for porosity and tortuosity of the mineral.

G, the mass fraction of the leachable copper sulphide mineral is given by the equation:

$$G = n_p \cdot A_p \cdot r_p \cdot \rho_p / 3\rho_r \qquad \text{(Eq. 6)}$$

where,

$n_p$ is the number of sulphide particles per unit volume of rock

$A_p$ is the average area per sulphide particle

$r_p$ is the average copper sulphide particle radius

$\rho_p$ is the density of copper sulphide

and $\rho_r$ is the bulk rock density.

It is useful to normalize the rate in terms of fraction reacted ($\alpha$), since $\alpha$ for a single particle or an assemblage of particle size $r_{io}$, is the same.

For a given particle:

$$1-\alpha = \frac{r_i^3}{r_{io}^3} = \frac{\text{volume of unreacted core}}{\text{volume of ore particle}} \qquad \text{(Eq. 7)}$$

The number of moles of unreacted mineral in the core of the particle is given by the expression:

$$n = \frac{4}{3} \cdot \frac{\pi r_i^3 \cdot \rho_r \cdot G}{M} \qquad \text{(Eq. 8)}$$

where,

M is the molecular weight of the copper sulphide mineral.

For a given value of $r_i$, equation 5 can be integrated for steady-state conditions ($J_i$ = constant) for all r values between $r_i$ and $r_{io}$, which upon combination with equation 4 gives an expression for the concentration of the reactant $C_i$ in the reaction zone in terms of the bulk solution concentration $C_b$ as:

$$C_i = C_b \left[ \frac{1}{1 + \left( \frac{\delta \cdot n_p \cdot A_p \cdot \sigma \cdot k}{Df} \right) \left( \frac{r_i}{r_{io}} \right) (r_{io} - r_i)} \right] \qquad \text{(Eq. 9)}$$

Substituting in equation 1 gives the generalized rate equation

$$\left[ \frac{dn}{dt} \right]_i = - \left[ \frac{4\pi r_i C_b}{\phi_{io}} \right] \left[ \frac{1}{\frac{1}{\delta \cdot n_p A_p k} + \left( \frac{\sigma}{Df} \right) \left( \frac{r_i}{r_{io}} \right) (r_{io} - r_i)} \right] \qquad \text{(Eq. 10)}$$

Combining equations (10) and (6) gives

$$\left[\frac{dn}{dt}\right]_i = -\left[\frac{4\pi r_i^2 C_b}{\phi_{io}}\right]\left[\frac{1}{\frac{1}{G\beta} + \left(\frac{\sigma_i}{Df\, r_{io}}\right)^{(r_{io}-r_i)}}\right] \qquad \text{(Eq. 11)}$$

where,

$$\beta = \frac{3\rho_r \cdot \rho \cdot k}{r_p \cdot \rho_p} \qquad \text{(Eq. 12)}$$

Combining equations (11) and (8) gives

$$\frac{dr_i}{dt} = -\frac{MC_b}{\phi_{io} \cdot \rho_r \cdot G}\left[\frac{1}{G\beta + \left(\frac{\sigma r_i}{Df\, r_{io}}\right)^{(r_{io}-r_i)}}\right] \qquad \text{(Eq. 13)}$$

Equations (7) and (13) may be combined to determine the fractional reaction rate $\frac{d\alpha}{dt}$ as:

$$\frac{d\alpha}{dt} = \frac{3MC_b}{\phi_{io} \cdot \rho_r \cdot G \cdot r_i}\left[\frac{(1-\alpha)^{2/3}}{\frac{1}{G\beta} + \left(\frac{\sigma \cdot r_{io}}{Df}\right)^{(1-\alpha)^{1/3}\{1-(1-\alpha)^{1/3}\}}}\right] \qquad \text{(Eq. 14)}$$

On integration equation (14) becomes

$$1 - \frac{2}{3}\alpha - (1-\alpha)^{2/3} + \frac{\beta'}{G \cdot r_{io}}[1 - (1-\alpha)^{1/3}] = \frac{\gamma'}{G \cdot r_{io}^2}\, t \qquad \text{(Eq. 15)}$$

where

$$\beta' = \frac{2Df\, r_p \rho_p}{3\sigma \rho_r \delta k} = \frac{2Df}{\sigma\beta} \qquad \text{(Eq. 16)}$$

and $$\gamma' = \frac{2MDfC_b}{\phi_{io} P_r \sigma} = \frac{\gamma}{\phi_{io}} \qquad \text{(Eq. 17)}$$

Equation (15) can be used to obtain copper extraction curves when the ore particle size range is narrow enough to be adequately described by one average size, but for sample with a broad distribution of particle sizes:

$$\alpha_{total} = \sum_j \alpha_j F_j \qquad \text{(Eq. 18)}$$

where,

$\alpha_j$ is the fraction reacted for size $r_{10}$

and $F_j$ is the fraction of the copper contained in size j.

The constants $\gamma'$ and $\beta'$ in equation (15) are dependent on the ore type and the reaction conditions present. As seen from equations (16) and (17), $\beta'$ depends on the ore type, while $\gamma'$ is dependent concentration of the reactant and $\phi_{io}$, which is the sphericity and roughness factor, in addition to the ore type.

## RESULTS

The results of the chemical analysis of the head sample ore, fines and a composite sample (head sample and fines mixed in the ratio in the original ore) are given in Table 3. The ore loaded into the columns had a copper content of 1.28%.

The immersion porosity of the rock was 0.45%. The porosity was only marginally higher than the average value of crystalline rocks (0.3-0.4).

The results of the mineralogical analyses are in Tables 4 to 6. The major constituents of the ore were quartz and chlorite, while pyrite, pyrrhotite and chalcopyrite were minor constituents. The ore also had trace quantities of siderite (Table 5).

Shake-flask tests give an indication as to the maximum amount that could be leached out in practice. The ore from Opemiska gave an extraction of 82% of the copper present.

The equilibration of the ore in the column took 30 days and needed 2.6 L of concentrated sulphuric acid (11 mg of acid for every kg of the ore). The average daily addition of acid during leaching was approximately 40 mL (0.18 mg per kg of the ore).

The results of the column leaching are given in Tables 7 and 8 and are plotted in Figure 4. The leaching rate is seen to increase with time and has accelerated to more than five times after a period of about 140 days. Average leaching rate for columns 3 and 4 are given below:

| Period (in days) | Leaching rate g/d Column 3 | Column 4 |
|---|---|---|
| 0-50 | 0.34 | 0.37 |
| 50-100 | 0.51 | 0.61 |
| 100-140 | 1.11 | 1.19 |
| 140-170 | - | 1.87 |
| 140-180 (Run #2) | 1.87 | - |

Extraction of 1.87 g/d gives a rate of 1.05% (of the copper present in the ore) per month. The extraction rate is still increasing and has not levelled off.

## DISCUSSION

The parameters that affect leaching by ferric sulphate (15) are:

(a) Surface area: The surface area is inversely proportional to the ore size. The cost of size reduction of the ore increases exponentially as the size reduction is continued to fine particles. The smallest size that could be blasted out of rocks economically appears to be around minus 15 cm minus 6 in.). Hence these columns were filled with minus 15 cm (minus 6 in.) ore.

(b) Temperature: The temperature coefficient for the dissolution rate is reported to be 1.25/10°C. The principal column leaching was carried out at room temperature. Since the temperature underground will not be different from the room temperature by more than 10°C it is assumed that the rate of leaching will not be very different from the values observed in the column.

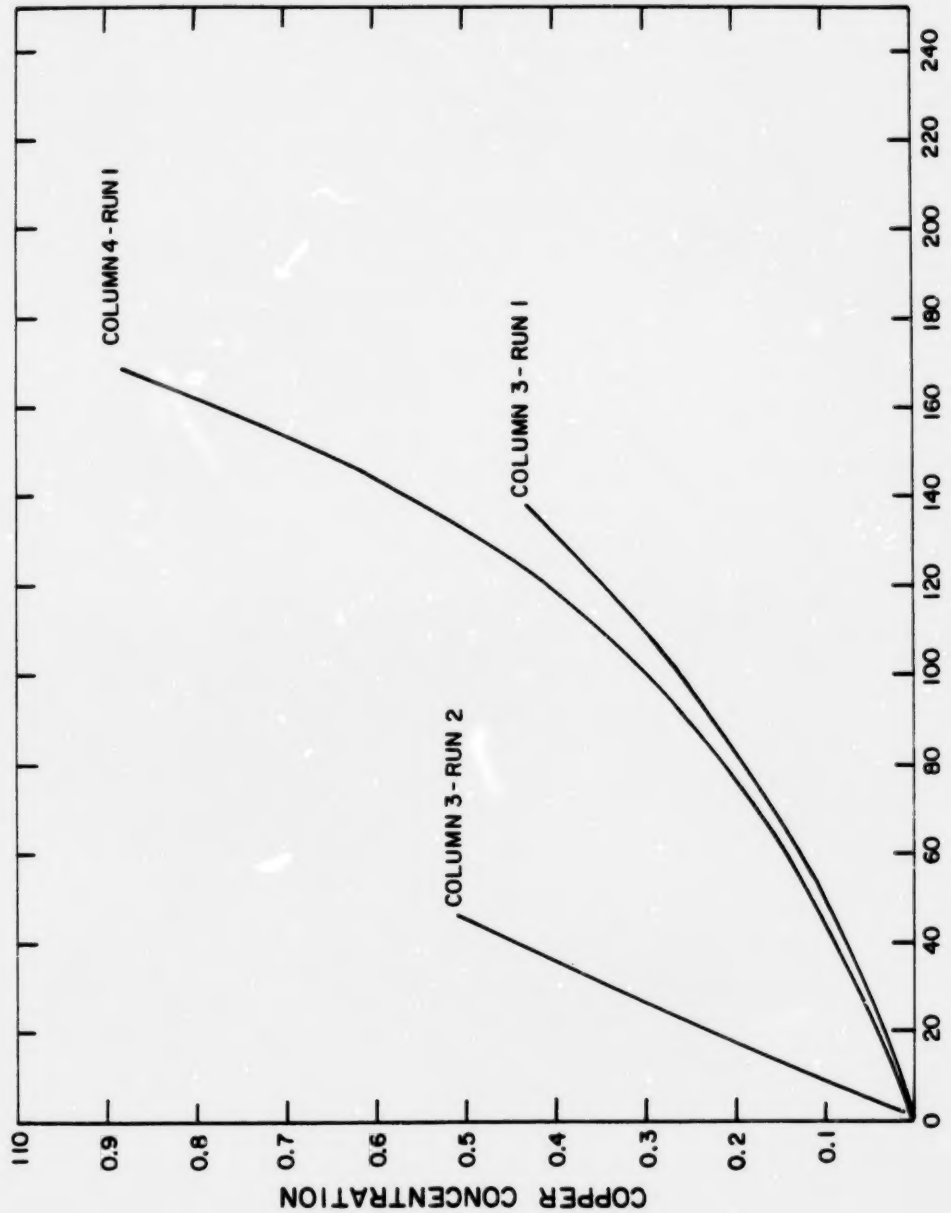



FIGURE 4 - Copper Extraction Curves for Column Leaching

(c) Copper content: Higher copper content in the ore will give better extraction rates. However, since this process is for waste rock and rocks that cannot be economically processed by conventional methods 1.28% is considered to be adequate.

(d) Porosity: Since the mechanism of ferric sulphate leaching is by diffusion a highly porous rock would be best for leaching. The rock from Opemiska had a porosity of only 0.45% (which is about the average value for crystalline rocks).

(e) Concentration of ferric sulphate: Small column tests conducted with zinc ore on the leach scale indicated that the extraction rate is independent of the ferric iron concentration. This conclusion is consistent with the findings of Dutrizac (16). The results of the present experiment also lead to the same conclusions, in that the extraction rates in columns 3 during the period 140-170 days and in column 4 in run 2 are the same although the ferric iron concentrations are different.

(f) Concentration of ferrous and copper sulphates: It has also been reported that the initial concentrations of ferrous and zinc sulphides had no effect on the extraction rate of zinc. It was expected that the situation would be similar in the case of copper leaching as well. The leaching rates obtained in column 3 during run 2 and in column 4 during the period 140-170 days confirm that the concentrations of ferrous and copper sulphates have no effect on copper extraction rates.

Models similar to the one derived under theoretical considerations have been used to predict pressure leaching of chalcopyrite in an autoclave (13), leaching of low-grade sulphide ores by percolating dilute sulphuric acid (12), and heap leaching of pyritic sulphide minerals (14).

Braun et al. (13) encountered the same phenomenon of accelerated leaching after 150 days. These authors concluded that the accelerated leaching is the result of an increase in the reaction surface. The effect is illustrated diagramatically in Fig. 5. In the case of the ore from Opemiska the presence of high iron which is solubilized by the ferric sulphate enhances the increase in the reaction surface.

The results obtained so far are inadequate to evaluate the parameters of the model.

Madsen and Groves of USBM (17) conducted percolation leaching of chalcopyrite-bearing ore at ambient and elevated temperatures with bacteria. The ore sizes used by them are very similar to the ores in this experiment. Six and four-tenths tonnes of a granitic ore obtained from Arizona copper mine were leached in fibreglass column 1.37 m in diameter and 3.04 m high. Maximum extraction rate reported was 0.44% per 100 days.

Scheffer and Evans reviewed copper leaching practices in the Western United States (18). According to these authors dump leaching of a chalcocite containing mineral gave an extraction rate of about 1.3% per month. (The calculations were made by the authors of this paper on the assumption that it contained 0.4% Cu.) These are results of the dump leaching operation of the American Smelting and Refining Co. in Arizona.

Lewis and Bhappu (19) in an economic evaluation of the available processes for treating oxide-copper ores, concluded that in-situ leaching of 1,650,000 tonnes per year containing 1% copper would cover all expenses (capital, operating, taxes etc.) in one year. This according to the calculations of the authors (of the present paper) is based on an extraction rate of 0.05% per year.

Thus it would appear that the extraction rate obtained with the Opemiska ore (after about 150 days of initial operation) is promising. However, it is suggested that the results of the Prefeasibility Analysis of Bacterially Assisted Underground Leaching of a Canadian Copper Sulphide Deposit study by Noranda Mines (DSS File No. 07SQ23440-4-9071) in which a sensitivity analysis and economic evaluation criteria will be detailed, be awaited before final conclusions are drawn.

## CONCLUSIONS

Shake flask leaching tests of finely ground ore gave 82% extraction. Column leaching of ore fractions ranging from +0.64 cm to -15.24 cm gave leaching rates which accelerated with time. The rate after 140 days was 1.1% per month and still increasing. The acid consumption during leaching was 0.18 mg of concentrate $H_2SO_4$ per kg of ore per day. The results

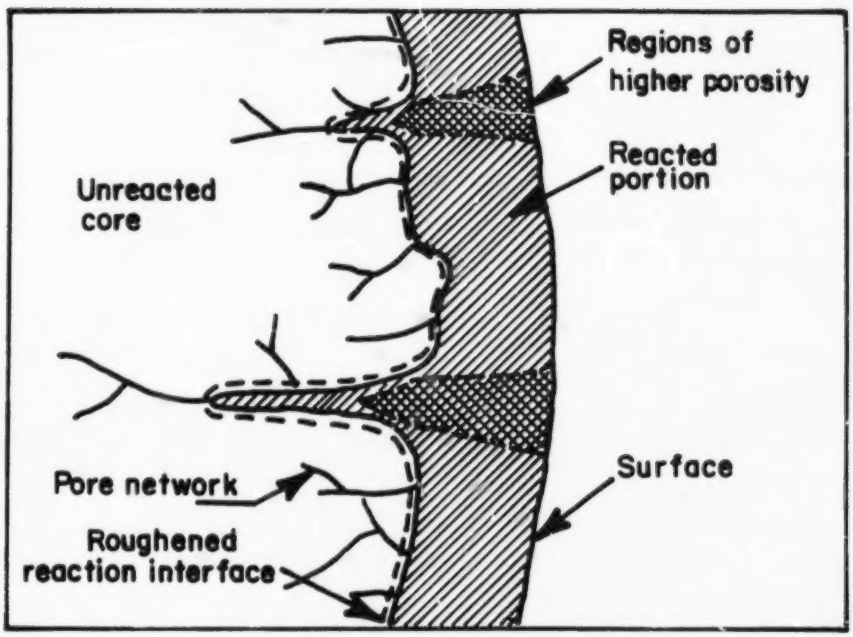



FIG. 5 - Partially Leached Ore Showing Roughening of Reaction Interface Caused by Preferential Leaching Through Mineralized Zones and Imperfections

( Reproduced from Braun et al. (12) )

obtained so far indicate that the Opemiska ore is a suitable candidate for in-place underground bacterially assisted two-stage leaching process. However, further work and an economic feasibility study needs to be conducted before a final decision is made.

## RECOMMENDATIONS AND FUTURE WORK

1. It is recommended that the column leaching be continued to determine when the extraction rate levels off and the maximum extraction rate that could be obtained in the columns.

2. If, after the results of the Noranda study are made available, it is found that the leaching rates obtained are economical, it is recommended that individual leaching curves for various size fractions be determined.

3. Finally an economic evaluation of the process should be done based on the mine site, facilities available and the quantity of ore available.

## ACKNOWLEDGEMENTS

The assistance of T.J. Katsube of G.S.C. in immersion porosity determinations is acknowledged. The contribution of other members of the Extraction Metallurgy Division to this work is acknowledged. The authors wish to thank the staff of the Chemical Laboratory and the Mineralogy Section of the Mineral Processing Laboratory for doing the various analyses.

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Table 1. Composition of modified 9 K medium

| | | |
|---|---|---|
| $(NH_4)_2SO_4$ | | 3.0 g |
| $K_2HPO_4$ | | 0.5 g |
| $MgSO_4.7H_2O$ | | 0.5 g |
| KCl | | 0.5 g |
| $Ca(NO_3)_2.4H_2O$ | | 0.01 g |
| Water | to | 1 L (Total Volume) |

Table 2. Size distribution of ore loaded into the columns

| Fraction | Weight of ore kg, (lb) | Percentage | Cumulative Percentage |
|---|---|---|---|
| -15.24 cm, +12.7 cm (-6", +5") | 5.78 (12.75) | 1.4 | 1.4 |
| -12.7 cm, +10.16 cm (-5, +4) | 26.08 (57.5) | 6.3 | 7.7 |
| -10.16 cm, +7.62 cm (-4, +3) | 80.17 (176.75) | 19.3 | 27.0 |
| -7.62 cm, +5.08 cm (-3, +2) | 137.21 (302.5) | 33.0 | 60.0 |
| -5.08 cm, +3.81 cm (-2, +1.5) | 68.49 (151.0) | 16.5 | 76.5 |
| -3.81 cm, +2.54 cm (-1.5, +1.25) | 36.63 (80.75) | 8.8 | 85.3 |
| -3.18 cm, +2.54 cm (-1.25, +1.0) | 34.02 (75.0) | 8.2 | 93.5 |
| -2.54 cm, +1.91 cm (-1.0, +0.75) | 15.56 (36.5) | 4.0 | 97.5 |
| -1.91 cm, +1.27 cm (-0.75, +0.50) | 6.35 (14.0) | 1.5 | 99.0 |
| -1.27 cm, +0.64 cm (-0.5, +0.25) | 4.88 (10.75) | 1.0 | 100.0 |
| Total | 917.5 (416.18) | | |

Table 3. Chemical analysis of head sample ore from Opemiska

| Element | Ore | Fines | Composite |
|---|---|---|---|
| Cu | 1.28 per cent | 2.08 per cent | 1.41 per cent |
| Zn | 0.05 per cent | 0.05 per cent | 0.05 per cent |
| Fe | 29.39 per cent | 29.4 per cent | 29.4 per cent |
| S | 16.41 per cent | 16.4 per cent | 16.4 per cent |
| C | 0.62 per cent | 0.6 per cent | 0.6 per cent |
| $SiO_2$ | 27.15 per cent | 27 per cent | 27 |
| Ni | 0.01 per cent | 0.01 per cent | 0.01 per cent |
| Pb | 0.03 per cent | 0.01 per cent | 0.03 per cent |
| Ag | 6.3 (ppm) | 12.6 (ppm) | 6.6 (ppm) |
| Au | 1.0 (ppm) | 4.9 (ppm) | 1.2 (ppm) |

Table 4. Size Distribution of Metallic Minerals

The data reported are the percentages of the surface area covered by the grain within each size range. The percentages are given as per cent smaller than cumulative per cent; and as in each size range (average). A 4X objective was used on the Zeiss microscope at 1.25. At this magnification 1 pp covers 2.20 $\mu m^2$.

Opemiska Ore

Chalcopyrite

| Size Micrometers | Total Counts | Difference Per cent | Cumulative Per cent |
|---|---|---|---|
| 37.1 | 757642 | 5.0 | 5.0 |
| 52.4 | 736598 | 1.6 | 6.6 |
| 74.0 | 707297 | 14.4 | 21.0 |
| 104.0 | 598170 | 11.0 | 32.0 |
| 147.0 | 511722 | 5.0 | 37.0 |
| 208.0 | 508436 | 5.0 | 42.0 |
| 294.0 | 438067 | 4.0 | 46.0 |
| 417.0 | 408544 | 7.0 | 53.0 |
| 489.0 | 281134 | 47.0 | 100.0 |
| 833.0 | 0 | 0 | 100.0 |

Pyrite

| Size Micrometers | Total Counts | Difference Per cent | Cumulative Per cent |
|---|---|---|---|
| 37.1 | 5674490 | 5.5 | 5.5 |
| 52.4 | 5363028 | 0.9 | 6.4 |
| 74.0 | 5311302 | 1.4 | 7.0 |
| 104.0 | 5175022 | 1.0 | 8.0 |
| 147.0 | 5158117 | 1.0 | 9.0 |
| 208.0 | 5139407 | 1.0 | 10.0 |
| 294.0 | 5130092 | 3.0 | 13.0 |
| 417.0 | 4921623 | 8.0 | 21.0 |
| 489.0 | 4500022 | 21.0 | 42.0 |
| 833.0 | 3282541 | 58.0 | 100.0 |

Pyrrhotite

| Size Micrometers | Total Counts | Difference Per cent | Cumulative Per cent |
|---|---|---|---|
| 37.1 | 2780055 | 4.5 | 4.5 |
| 52.4 | 2655255 | 3.3 | 7.8 |
| 74.0 | 2564171 | 0.2 | 8.0 |
| 104.0 | 2559328 | 5.0 | 13.0 |
| 147.0 | 2408130 | 3.0 | 16.0 |
| 208.0 | 2326412 | 9.0 | 25.0 |
| 294.0 | 2075351 | 8.0 | 33.0 |
| 417.0 | 1865856 | 3.0 | 39.0 |
| 489.0 | 1695898 | 37.0 | 76.0 |
| 833.0 | 673891 | 24.0 | 100.0 |

Table 5. X-ray diffractometer results

Lake Opemiska

Major-quartz, chlorite
Minor-pyrite, pyrrhotite, chalcopyrite
Trace-siderite

Table 6. Calculated weight percentages from image analyses*

Lake Opemiska

| | |
|---|---|
| gangue** | 77.5 |
| pyrite | 14.5 |
| pyrrhotite | 6.4 |
| chalcopyrite | 1.6 |

*Assuming a special gravity of 3.0 for gangue, 5.0 for pyrite, 4.5 for pyrrhotite, 4.2 for chalcopyrite and 5.0 for Fe oxides

**Gangue = quartz, chlorite with a trace of siderite

Table 7. Results of column leaching
Run no. 1

| Day | Column 3 (Stationary) | | | Day | Column 4 (Circulating) | | |
|---|---|---|---|---|---|---|---|
| | Total Fe g/L | $Fe^{2+}$ g/L | Cu g/L | | Total Fe g/L | $Fe^{2+}$ g/L | Cu g/L |
| 1 | 4.47 | 0.61 | | 1 | 5.0 | 0.004 | 0.002 |
| 7 | 2.90 | 0.004 | 0.006 | 6 | 5.33 | 0.001 | 0.015 |
| 14 | 4.57 | 0.87 | 0.030 | 12 | 5.52 | 1.16 | 0.028 |
| 20 | 4.91 | 1.56 | 0.041 | 18 | 5.53 | 1.39 | 0.028 |
| 26 | 5.37 | 1.53 | 0.045 | 24 | 5.95 | 1.61 | 0.042 |
| 33 | 5.68 | 2.32 | 0.062 | 31 | 6.72 | 2.03 | 0.064 |
| 40 | 5.59 | 2.12 | 0.079 | 37 | 7.50 | 3.03 | 0.083 |
| 47 | 6.03 | 2.64 | 0.093 | 44 | 8.87 | 3.97 | 0.018 |
| 54 | 6.28 | 3.14 | 0.11 | 51 | 9.74 | 4.76 | 0.12 |
| 61 | 6.68 | 3.32 | 0.13 | 58 | 9.85 | 5.39 | 0.15 |
| 68 | 6.94 | 3.48 | 0.15 | 65 | 10.4 | 5.69 | 0.17 |
| 75 | 6.92 | 3.95 | 0.17 | 72 | 10.8 | 6.15 | 0.20 |
| 82 | 7.67 | 3.84 | 0.20 | 79 | 10.6 | 6.20 | 0.22 |
| 89 | 7.53 | 4.28 | 0.22 | 86 | 10.5 | 5.92 | 0.23 |
| 96 | 8.37 | 4.60 | 0.25 | 93 | 11.1 | 6.51 | 0.27 |
| 103 | 8.21 | 4.89 | 0.27 | 100 | 7.4 | 6.20 | 0.31 |
| 110 | 5.20 | 2.09 | 0.28 | 107 | 7.4 | 7.29 | 0.34 |
| 117 | 5.61 | 4.90 | 0.31 | 114 | 11.9 | 7.35 | 0.41 |
| 124 | 8.82 | 5.05 | 0.36 | 121 | 12.3 | 8.14 | 0.47 |
| 132 | 8.83 | 5.56 | 0.42 | 127 | 12.7 | 7.58 | 0.51 |
| 138 | 8.56 | 5.20 | 0.44 | 134 | 12.4 | 5.14 | 0.55 |
| | | | | | CIRCULATION STOPPED | | |
| | | | | 141 | 12.5 | 7.79 | 0.59 |
| | | | | 149 | 13.6 | 9.36 | 0.66 |
| | | | | 155 | 13.2 | 9.6 | 0.72 |
| | | | | 162 | 14.9 | 10.4 | 0.79 |
| | | | | 169 | 15.2 | 11.7 | 0.88 |
| | | | | 176 | 15.2 | 12.0 | 0.89 |

Table 8. Results of column leaching stationary column (column 3)

Run No. 2

| | Total Fe g/L | $Fe^{2+}$ g/L | Cu g/L |
|---|---|---|---|
| 1 | 9.47 | 0.28 | 0.01 |
| 7 | 9.38 | 1.35 | 0.10 |
| 15 | 9.71 | 2.22 | 0.17 |
| 21 | 10.2 | 2.54 | 0.21 |
| 28 | 10.9 | 3.22 | 0.30 |
| 35 | 11.1 | 4.40 | 0.39 |
| 42 | 11.8 | 5.04 | 0.50 |